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Process for dissolving precious metals

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ABSTRACT

A process for dissolving precious metals, particularly trace amounts of gold, platinum, palladium, rhodium, ruthenium, osmium and iridium found in mineral ores and concentrates.

- 5 A slurry comprising a precious metal bearing material and an aqueous acidic solution is heated to a temperature of at least 150°C in the presence of a complexing agent and a non-halogen oxidising agent. The complexing agent is selected to form soluble complexes of the precious metals, such that the precious metals are leached into solution during dissolution.

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INVENTION TITLE: "PROCESS FOR DISSOLVING PRECIOUS
METALS"

The following Statement is a full description of this invention including the best method of performing it known to me/us:

TITLE

"PROCESS FOR DISSOLVING PRECIOUS METALS"

5

FIELD OF THE INVENTION

The present invention relates to a process for dissolving precious metals, particularly precious metals present in trace amounts, from mineral ores and concentrates. The term precious metals in this application should be understood to include also minerals
10 formed from such metals.

BACKGROUND OF THE INVENTION

15 It is well established in hydrometallurgical mineral processing that a range of leaching agents are available to facilitate dissolution of first row transition metals such as cobalt, nickel, or copper, from mineral ores and concentrates. The efficacy of most leaching agents relies on the oxidation and/or selective complexation of metal cations in solution, sometimes coupled with an ability of the leaching agent to lower
20 the oxidation potential of the metal such that potential energy savings in the hydrometallurgical process is possible compared with more traditional pyrometallurgical processing.

The precious metals are considered to be "noble" metals, and include gold, platinum, palladium, rhodium, ruthenium, osmium and iridium. They normally remain inert to
25 oxidation and dissolution under ambient or high temperature and pressure conditions within acidic and/or oxidising leaching solutions. Furthermore, trace amounts of precious metals may occur with, or be occluded by base metal ores in the mineral matrix, thereby preventing physical contact between the leaching agent and the precious metals, and inhibiting the reaction therebetween. Thus, under the conditions

of known hydrometallurgical processes, trace amounts of precious metals commonly reside in the leach residues and a separate leaching process is generally required in order for the precious metals to be recovered.

It is known to provide methods for the dissolution of precious metals from materials
5 containing said metals. Hydrometallurgical recovery of precious metals has traditionally been achieved by extraction with cyanides, aqua regia, and more recently by oxidative treatment with elemental halogens. Such materials are not only expensive but because of their toxicity, particularly in the case of cyanide, expensive transportation, storage and environmental protection facilities are required for their
10 use in a processing installation.

Furthermore, it is common practice for some precious metal containing ores particularly sulphidic ores to be subjected to high temperature roasting of the ore prior to hydrometallurgical leaching, to liberate precious metals occluded by the mineral matrix.

15 The present invention seeks to overcome at least some of the aforementioned disadvantages.

SUMMARY OF THE INVENTION

In accordance with one aspect of the present invention there is provided a method for
20 the dissolution of a precious metal, including heating a slurry to a temperature of at least 150°C in the presence of a complexing agent and a non-halogen oxidising agent, the complexing agent being selected to form soluble complexes of the precious metals leached into solution during dissolution, wherein the slurry comprises a precious metal bearing material and an aqueous acidic solution, and the slurry is maintained at
25 a pH of less than 4 during dissolution.

DETAILED DESCRIPTION OF THE PREFERRED EMBODIMENT

The process of the present invention is suitable for the dissolution of precious metals present in trace amounts in solid materials, such as mineral ores and concentrates, wherein the precious metal is in an elemental form, a metal oxide form, a metal sulphide form, or other form. It will be understood that the mineral ore or concentrate may also contain one or more base metals, for example copper, cobalt, nickel, or zinc. It will also be understood that the mineral ore or concentrate may include metal sulphides, metal oxides or other minerals formed by the precious metals.

Preferably, the mineral ore or concentrate may be first ground to form a fine particulate solid of approximately 200 micron or less. In this way, the surface area of each particulate is greatly increased to allow optimum exposure to the leaching chemicals. Preferably, the fine particulate solid is slurried in aqueous solution during comminution or alternatively mixed with an aqueous solution after comminution to form a slurry which may contain from 5 to 50 weight percent solids in water.

A complexing agent is introduced or mixed with the slurry. Preferably, the complexing agent is a halide ion, such as chloride, bromide, iodide, fluoride or a mixture thereof. Typically, at least a sufficient amount of the complexing agent is added to the slurry to form soluble complexes of the precious metals leached into solution during the dissolution process.

The slurry may then be transferred to a pressure vessel or autoclave.

The slurry in the pressure vessel or autoclave may then be heated to a temperature of at least 150°C, preferably between 200°C to 250°C. Preferably, the acid is then added to the slurry. Preferably, the acid is a strong inorganic acid, such as sulphuric acid, nitric acid, an aqueous hydrogen halide or a mixture thereof.

The slurry is treated under the above mentioned conditions with a non-halogen oxidising agent. Preferably, the non-halogen oxidising agent is oxygen gas, which may be supplied to the pressure vessel or autoclave at an overpressure from about 50 kPa to 500 kPa. Alternatively, the non-halogen oxidising agent may be hydrogen peroxide, nitric acid, peroxymonosulphate salts, manganese oxides, chlorate salts, or a mixture thereof. It will be appreciated that non-gaseous non-halogen oxidising agents are typically mixed into the slurry, or in the case of a gas, either dispersed into the slurry and/or applied to the atmosphere above the slurry, after the slurry is transferred to the pressure vessel or autoclave.

10 Preferably, the dissolution process may be conducted over a period from about 30 minutes to 2 hours. Preferably, the final pH of the slurry is less than 4.0.

Under the reaction conditions described above, both base metals and precious metals are dissolved into solution. The precious metals can be separated from the base metals and removed from the solution via conventional techniques.

15 For lateritic ores or concentrates, the primary role of the inorganic acid is to dissolve the base metal oxides into solution to facilitate contact between the precious metals, which are occluded by the base metal oxides in the mineral matrix, and the non-halogen oxidising agent. The inorganic acid also provides a lower pH medium in which the dissolved metal ions and metal ion complexes are more likely to be soluble.

20 In sulphide ores or concentrates, the precious metals may occur with, or are occluded by base metal sulphides in the mineral matrix. The non-halogen oxidising agent oxidises insoluble sulphide species to soluble higher oxidation state sulphur species, such as sulphate. The non-halogen oxidising agent also oxidises the precious metals to a higher oxidation state ionic species which are conducive to forming soluble precious
25 metal complexes with the complexing agent.

The advantage of the present invention is that the precious metals as well as the other base metals present in the mineral matrix are dissolved into solution in the same unit process. The various metals can then be separated and recovered from solution by well established processes.

5 The following example illustrates the invention:

Example 1

A laterite ore, containing nickel, cobalt and platinum, was screened at 212 micron, and the oversize crushed to less than 212 micron.

10 An ore slurry formed from the crushed laterite ore (600g) and a saline solution (1800g) of 2.5 M NaCl and 0.4 M MgSO₄ was placed in an autoclave vessel. The autoclave vessel was sealed and heated to 250°C. At this temperature, the steam pressure was 4,800 kPa. An oxygen gas overpressure of 400 kPa was supplied to the sealed autoclave vessel and maintained throughout the dissolution reaction. Concentrated sulphuric acid (300g) was then added to the sealed autoclave vessel
15 through an inlet valve and the reaction was allowed to proceed for 2 hours. The resultant slurry was then cooled to below 100°C, and a representative sample was taken for analysis. The supernatant was filtered from the solid.

20 Analytical test results were as follows. The final pH of the solution was 0.6. The oxidation/reduction potential of the filtrate was 800 mV (vs Ag/AgCl). The final concentration of H₂SO₄ was 35 g/L. The recovery values of nickel, cobalt and platinum after dissolution under the conditions specified above were 95%, 97% and 75% respectively.

Example 2

25 A laterite ore, containing nickel, cobalt, gold, platinum and palladium together with high concentrations of manganese oxides, was screened at 212 micron, and the

oversize was ground to less than 212 micron. The concentrations of nickel, cobalt and manganese in the ore were 1.2%, 0.11%, and 5.2%, respectively.

An ore slurry formed from the crushed laterite ore (600g) and a saline solution (1800g) of 1.0 M NaCl was placed in an autoclave vessel. The autoclave vessel was
5 sealed and heated to 250°C. At this temperature, the steam pressure was 4,800 kPa. In contrast to Example 1, an overpressure of oxygen was not supplied to the vessel. Sufficient manganese oxide was present in the laterite ore to act as the non-halogen oxidising agent.

Concentrated sulphuric acid (220g) was then added to the sealed autoclave vessel
10 through an inlet valve and the reaction was allowed to proceed for 2 hours. The resultant slurry was then cooled to below 100°C, and a representative sample was taken for analysis. The supernatant was filtered from the solid.

Analytical test results were as follows. The oxidation/reduction potential of the filtrate was 810 mV (vs Ag/AgCl). The final concentration of H₂SO₄ was 40 g/L.

15 The recovery values of nickel, cobalt, gold, platinum and palladium after dissolution under the conditions specified above were 96%, 97%, 80%, 85%, and 90%, respectively.

Example 3

A nickel refinery waste product, containing significant metal values of nickel, cobalt
20 and platinum group metals, was mixed with a nickel/cobalt laterite ore containing high concentrations of manganese oxides. The mass ratio of the waste product to the laterite ore was 2:1.

The concentrations of nickel, cobalt and manganese in the ore were 1.4%, 0.52%, and 15% respectively. The refinery waste contained 3.0% nickel, 0.3% cobalt, 4 ppm
25 gold, 200 ppm platinum, and 150 ppm palladium.

The mixture of waste product and laterite ore was screened at 212 micron, and the oversize was crushed to less than 212 micron.

An ore slurry formed from the crushed mixture (600g) and a saline solution (1800g) of 0.5 M NaCl was placed in an autoclave vessel. The autoclave vessel was sealed and heated to 250°C. At this temperature, the steam pressure was 4,800 kPa.

In contrast to Example 1, an overpressure of oxygen was not supplied to the vessel. Sufficient manganese oxide is present in the mixture to act as the non-halogen oxidising agent.

Concentrated sulphuric acid (250g) was then added to the sealed autoclave vessel through an inlet valve and the reaction was allowed to proceed for 2 hours. The resultant slurry was then cooled to below 100°C, and a representative sample was taken for analysis. The supernatant was filtered from the solid.

Analytical test results were as follows. The oxidation/reduction potential of the filtrate was 790 mV (vs Ag/AgCl). The final concentration of H₂SO₄ was 50 g/L.

The recovery values of nickel, cobalt, gold, platinum, and palladium after dissolution under the conditions specified above were 97%, 97%, 85%, 90%, and 90%, respectively.

Example 4

A manganese oxide ore, containing approximately 25% manganese, was mixed with a nickel/cobalt laterite in a mass ratio of three parts laterite ore to one part manganese oxide ore. The concentrations of nickel, cobalt and platinum in the ore were 1.2%, 0.12%, and 0.2 ppm respectively. The manganese oxide ore contained only minuscule quantities of these metals.

The mixed ore was screened at 212 micron, and the oversize was ground to less than 212 micron. An ore slurry formed from the crushed ore mixture (600g) and a saline

solution (1800g) of 0.2 M NaCl was placed in an autoclave vessel. The autoclave vessel was sealed and heated to 250°C. At this temperature the steam pressure was 4800 kPa. An overpressure of oxygen was not supplied to the vessel as sufficient manganese oxide was present in the laterite ore to act as the non-halogen oxidising agent.

Concentrated sulphuric acid (270g) was then added to the sealed autoclave vessel through an inlet valve and the reaction was allowed to proceed for 2 hours. The resultant slurry was then cooled to below 100°C, and a representative sample was taken for analysis. The supernatant was filtered from the solid.

Analytical test results were as follows. The oxidation/reduction potential of the filtrate was 800 mV (vs Ag/AgCl). The final concentration of H₂SO₄ was 50 g/L. The recovery values of nickel, cobalt, and platinum after dissolution under the conditions specified above were 95%, 95% and 90% respectively.

While the present invention has been particularly described with reference to certain specific embodiments thereof it will be understood that various modifications may be made to the process by persons skilled in the art without departing from the spirit and scope of the invention.

CLAIMS

1. A method for the dissolution of a precious metal, including heating a slurry to a temperature of at least 150°C in the presence of a complexing agent and a non-halogen oxidising agent, the complexing agent being selected to form soluble complexes of the precious metals leached into solution during dissolution, wherein the slurry comprises a precious metal bearing material and an aqueous acidic solution, and the slurry is maintained at a pH of less than 4 during dissolution.
2. The method according to claim 1, characterised in that the complexing agent is a halide ion selected from a halide ion group including chloride, bromide, iodide, fluoride or a mixture thereof.
3. The method according to claim 1 or claim 2, characterised in that the slurry is heated to a temperature between 150°C to 300°C.
4. The method according to claim 3, characterised in that the slurry is heated to a temperature between 200°C to 250°C.
5. The method according to any one of claims 1 to 3, characterised in that the aqueous acidic solution is comprised of a strong inorganic acid.
6. The method according to claim 5, characterised in that the strong inorganic acid is selected from a group including sulphuric acid, nitric acid, an aqueous hydrogen halide, or a mixture thereof.
7. The method according to any one of claims 1 to 6, characterised in that the non-halogen oxidising agent is oxygen.
8. The method according to claim 7, characterised in that oxygen is supplied at an overpressure from about 50kPa to 500kPa.

9. The method according to any one of claims 1 to 6, characterised in that the non-halogen oxidising agent is selected from a group including hydrogen peroxide, nitric acid, peroxymonosulphate salts, manganese oxides, chlorate salts, or a mixture thereof.

5 10. The method according to any one of claims 1 to 9, characterised in that the precious metal bearing material is a fine particulate solid screened in a range of 150 to 300 micron.

11. The method according to claim 10, characterised in that the precious metal bearing material is a fine particulate solid less than about 200 micron.

10 12. The method according to any one of claims 1 to 11, characterised in that the slurries contains 5% to 50% w/w solids.

13. The method according to any one of claims 1 to 12, characterised in that the precious metal bearing material is a sulphide ore or concentrate.

14. The method according to any one of claims 1 to 12, characterised in that the precious metal bearing material is an oxide ore or concentrate.

15. The method according to any one of claims 1 to 12, characterised in that the precious metal bearing material is a laterite ore or concentrate.

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